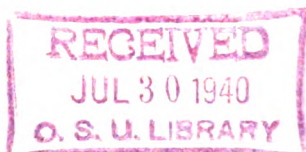


UNITED STATES
DEPARTMENT OF THE INTERIOR
HAROLD L. ICKES, SECRETARY

BUREAU OF MINES
R. R. SAYERS, ACTING DIRECTOR

INFORMATION CIRCULAR

MINING AND MILLING METHODS AND COSTS AT
KNOB HILL MINE, REPUBLIC, WASH.



BY

CORWIN L. COOPER

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO
THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

INFORMATION CIRCULAR

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

MINING AND MILLING METHODS AND COSTS AT
KNOB HILL MINE, REPUBLIC, WASH.^{1/}

By Corwin L. Cooper.^{2/}

INTRODUCTION

This paper, which describes mining and milling methods and costs at the Knob Hill gold mine, is one of a series being prepared by the Bureau of Mines on gold mining and milling practices in the United States.

The mining properties owned or controlled by Knob Hill Mines, Inc., are situated in the Eureka mining district on a country road 2 miles up Eureka Gulch from the town of Republic, county seat of Ferry County, Wash. The district is about 140 miles northwest of Spokane by road and 35 miles south of the international boundary line. The altitude at the mine office is 2,700 feet.

Good improved roads lead south and west from Republic, while a limited freight service to Republic is maintained by the Great Northern Railroad.

The district reposes on the eastern slopes of the great Cascade Range; summers are pleasant, though somewhat short; frosts come early in the fall and continue late in the spring. Winters are seldom severe, and company records taken regularly from indicating maximum and minimum thermometers, disclose that a mean temperature of plus 27° F. is customary for the 2 winter months. During the 2 years of operation there have been extreme instances where the temperature dropped to minus 15°, and historical data show lows far beyond this for short periods. Snowfall ranges from 4 to 7 feet, with an approximate maximum of 30 inches of snow on the ground at any one time.

Acknowledgment is made to the executives of the Knob Hill Mines, Inc., for permission to publish this paper, and to F. H. Mitchell, general superintendent, and L. A. Fitzgerald, mill superintendent, for their assistance in its preparation.

Water for company operations is pumped 3 1/2 miles from the San Poil River, a small stream skirting the town of Republic. Although the company has experienced no shortage of water, it is said that after seasons of extreme dryness, a scarcity of water may be anticipated.

- 1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Information Circular 7123."
- 2/ One of the consulting engineers, Bureau of Mines, and chief metallurgist, Knob Hill Mines, Inc.

The district is served with electric power by a local company, which purchases current from the Washington Water Power System and transmits it over 40 miles of line at a potential of 33,000 volts.

The district apparently became active in 1896 and seems to have reached the zenith of its discovery and exploratory activities by the turn of the century. As the gold was deposited in a finely disseminated state the ores were not readily amenable to the milling practice of the times, and intensive development and expansion were discouraged. Nevertheless, the operators of this district were among the first to use the cyanide process in this country, and recent investigations have added little to their metallurgical knowledge concerning these ores. Failure to treat these ores successfully in early mills probably was due first, to lack of modern, reliable, efficient equipment and, second, to inability of the mines to yield ore of a profitable grade in adequate daily quantities to meet the demands of the plants.

By 1913 twice as much ore was being shipped to the smelters as was being milled locally, and from that time the camp gradually declined until all activity ceased except desultory leasing. This was virtually the situation when the Knob Hill Mines, Inc., began operations in 1936.

GEOLOGY

The general geology of the Republic district has been described by Lindgren and Bancroft.^{3/} Development in recent workings has added further light on the subject.

A series of andesitic flows about 1,500 feet thick is overlain with lacustrine deposits of clay and sandy shales intercalated with later flows of more basic character than the earlier extrusions. Throughout most of the district, the whole series, including sediments, is tilted to the east at an average angle of 35°. Tertiary movement of considerable magnitude preceding vein emplacement has taken place, and post-mineral cross-faulting and vein brecciation cut the ore and tilted the host rocks. A period of erosion and glaciation lowered the surface an estimated 1,000 feet; this was followed by widespread extrusion of basaltic lavas in late Tertiary times. The veins belong to the Tertiary age.

The mineralized zone is about 6 miles long. The veins dip steeply eastward at the south end, becoming flatter toward the Knob Hill mines, and reverse their dip farther north at the Mountain Lion mine. The veins which are several hundred feet in length, have a maximum width of more than 20 feet.

The ore consists of milky to smoky quartz alternated with dark bands composed in part of very fine sulfides with which the gold and silver minerals are associated. Free gold is found rarely in the district, and none has been observed in the Mud Lake vein now being mined. Mainly, the

^{3/} Lindgren, Waldemar, and Bancroft, Howland: The Ore Deposits of Northwestern Washington. Republic Mining District: Geol. Survey Bull. 550, 1914, pp. 133-166.

precious metals seem to be associated with selenium, possibly varying in character from a normal selenide to a complex mixture. According to Lindgren and Bancroft, the mixture may contain not only gold and silver but also substantial percentages of antimony, sulfur and copper and small amounts of zinc, iron, arsenic, and selenium. At some points along the hanging wall of the Mud Lake vein, stibnite or some other complex sulfide of antimony seriously interferes with proper functioning of the mill. The presence of stibnite usually indicates a marked increase in the gold content of the ore. Some work done on the Mud Lake vein by other operators several years ago suggested the possibility of such a marked increase in the stibnite content at depth as to require an entirely different metallurgical treatment. Current work on the ores of the deeper levels in mines at the southern end of the district does not support this presumption.

The ratio between silver and gold in the ore of the Mud Lake vein has increased gradually as mining was extended into the hill, until at present it is about 6 to 1. This may be due to leaching of the silver near the surface.

MINING

The ores currently mined and milled by the company come from the Knob Hill and the Mountain Lion mines. As the mining methods are entirely different at each mine, they will be considered separately. The Knob Hill mine, being the chief source of ore and the control unit around which all the operations revolve, will be described in more detail than the Mountain Lion, which supplies only about 15 percent of the ore, as at present, it is virtually a development operation.

The mill, local office, assay office, general shops, and warehouses are at the Knob Hill mine.

Knob Hill Mine

Present mining is confined to the Mud Lake vein. This vein lies to the northwest, almost contiguous to the Knob Hill vein, which outcrops boldly, forming a prominence that gives the hill and mine their name. It was the source of ore mined in the past.

The Mud Lake vein outcrops but its full length is not yet known. It has been developed and is being mined by an open-cut for about 650 feet south from the outcrop; the bottom of the pit is now about 100 feet below the highest point on the outcrop. The pit will be worked about 50 feet deeper, which appears to be the economic limit for shovel operation.

Although the presence of the Mud Lake vein was known to earlier owners of the property, it could not be exploited profitably at the then prevailing prices for gold and silver. With the rise in the price of gold, the vein was prospected extensively by drifting, crosscutting, and diamond drilling. An inclined prospect shaft was sunk to a point 100 feet below the

present pit bottom. This shaft is now used as a sump to drain the floor of the pit. This development work, which was done before the property was acquired by the present operators, disclosed a body of ore 30 to 60 feet wide. It consists of quartz bands or veins that dip about 40° to the east through andesitic porphyry. The capping or overburden consists of carbonaceous clay shales and slates with areas of pyroxene andesite, all of which can be removed readily. The ratio of stripping to ore ranges from virtually nothing to nearly 3 to 1 at the lowest horizon of profitable pit operation. The shale hanging wall is well-defined and limits ore in that direction. The foot wall is not defined so clearly and in practice it is really a plane in the vein matter below which the ore grade becomes too low for profitable mining.

After due consideration, it was decided to mine the body with power shovels. A combination of shovels, caterpillars equipped with bulldozer blades and carry-alls, and trucks was selected for removing the overburden.

The proved tonnage and value of the Mud Lake vein originally was calculated from the exploratory work that has been described briefly. For control of mining, samples are taken of the cuttings from 16-foot holes drilled for blasting. The samples are assayed and mapped ahead of actual mining as a guide, to avoid getting too deep into the foot wall and in areas of lean value.

The program of prospecting has been laid out to determine the extension of the Mud Lake vein east of the Knob Hill vein. This program will consist of crosscutting and drifting from the old workings in the Knob Hill vein, supplemented by drilling from the surface and stripping of the top of the vein. The object of this work is to determine the southerly limits that may be mined by shovels, taking the depth of overburden and grade of ore into consideration. This prospecting also will determine the feasibility of using other mining methods where the shovels cannot be operated profitably.

The pit is now producing about 500 tons of ore daily, which requires removal of about 2 tons of overburden for each ton of ore mined. As the hanging wall consists of shales or slates, it is possible to keep the ore fairly free of waste in that direction. The distance broken into the foot wall is mainly a matter of maintaining a satisfactory grade of ore for the mill.

In general, the plan of mining is to keep the stripping operation well ahead of ore removal. As the pit has advanced southward into the hill, stripping has come to include removal of 5 to 10 feet of soil directly over the ore; this soil is a clayey material very difficult to strip in winter or during spring thaws. An ample supply of ore is uncovered in summer, when stripping can be done economically.

Mining is done by two 1 1/4-yard, gasoline-driven shovels equipped with caterpillar treads. The shovels are used for mining ore from either or both of two benches or for stripping waste. Ore is delivered to the mill by

5-ton dump trucks; there are six of these trucks, and one of them is kept in reserve or undergoing service or repairs. Since the haul ranges from less than 1,500 feet, when mining is from the extreme south end of the known ore, to more than 4,000 feet, when delivery is made from the pit itself, the number of trucks needed to haul the required daily tonnage to the mill ranges from three to five. Trucks not needed immediately for ore haulage generally are used to assist in stripping operations. The road system has been designed so that there is virtually no up-grade haul for loaded trucks except for the relatively short climb from the pit bottom.

Stripping is done mainly with a 12-yard Le Tourneau carry-all scraper drawn by an R. D. 8 Caterpillar tractor. The work progresses with excavation of benches, which are taken down in a series of steps. The ground is broken for the carry-all by an R. D. 7 Caterpillar tractor equipped with a "rooter" and a "bulldozer" blade. Occasionally, it is necessary to transport overburden for distances greater than the economic haul for the carry-all equipment. Such material is moved by the shovels and trucks on the night shift; all ore is delivered to the mill during day shift. It is now necessary to provide space surrounding the mine for spoil dumps, but when the north end of the pit reaches its ultimate depth it will be utilized for the disposal of waste. Solid rock in the foot wall stands well even when walls are vertical, but the slates and shales in the hanging wall soon slough when cuts are steeply inclined.

Ore is mined from benches at right angles to the strike of the vein. The faces are carried 16 feet high, so they may be broken to the bench grade with vertical holes, thus avoiding toe holes or lifters. At present all drilling is done with medium-weight jack hammers using up to 16-foot lengths of 7/8-inch hexagonal steel and Carr-type side-hole jack bits. Machines are operated at an air pressure of 100 pounds per square inch. A modern compressor with a capacity of 700 cubic feet of free air per minute is driven through V-belts by a 100 h.p. electric motor.

Drill holes are spaced 10 to 12 feet apart and are staggered to improve breaking. Holes are sprung and loaded with quarry-type low-strength bag powder. Charges are detonated with No. 6 electric caps with 20-foot wires; primers are placed in the bottom of the holes. Column loading is being tried in an effort to obtain better fragmentation and to reduce the amount of secondary blasting. Large pieces of ore are blasted by plugging and loading with stump blasting powder and detonating with cap and fuse. Powder consumption, including all uses, whether on ore or waste, is about 1/2 pound per ton of ore. Drilling is done by the day crew, which also does the blasting either during the lunch hour or at quitting time.

For night work all equipment is provided with electric headlamps supplemented by two 1,000-watt portable floodlights. Each of these floodlights is supplied with long lengths of heavy-duty, outdoor-type, portable cord attachable to outlets from a 120-volt circuit carried parallel to the pit.

Ore is recovered with virtually no dilution by waste. The hanging wall is sharply defined, and no difficulty is experienced in stripping it clean without sacrificing ore or diluting values, but careful and continuous sampling is necessary to avoid mining unprofitable sections of the foot wall.

A pit boss has direct charge of the mine. The day shift comprises about 20 men, and the night crew consists of about 10 men, including the night foreman. The wage scale is comparatively high and is above that prevailing in the district. Particular attention is paid to maintenance of safe working conditions; the banks of the pit represent a constant hazard and must be scaled carefully and kept free of loose rock. The lost-time accident record is remarkably good, with but one serious injury recorded for the entire operation of the pit.

Mountain Lion Mine

The Mountain Lion mine, which is situated about 1 mile northwest of the Mud Lake pit, is owned and operated by a separate organization, although actually controlled by Knob Hill Mines, Inc. The vein probably was among the first discovered in the district, but records of production are incomplete and unreliable. At one time there was a stamp mill at the mine, which later was supplemented by a sand-leaching plant using the cyanide process. In recent years only a small tonnage of ore has been produced, mainly by lessors, who shipped directly to the smelters.

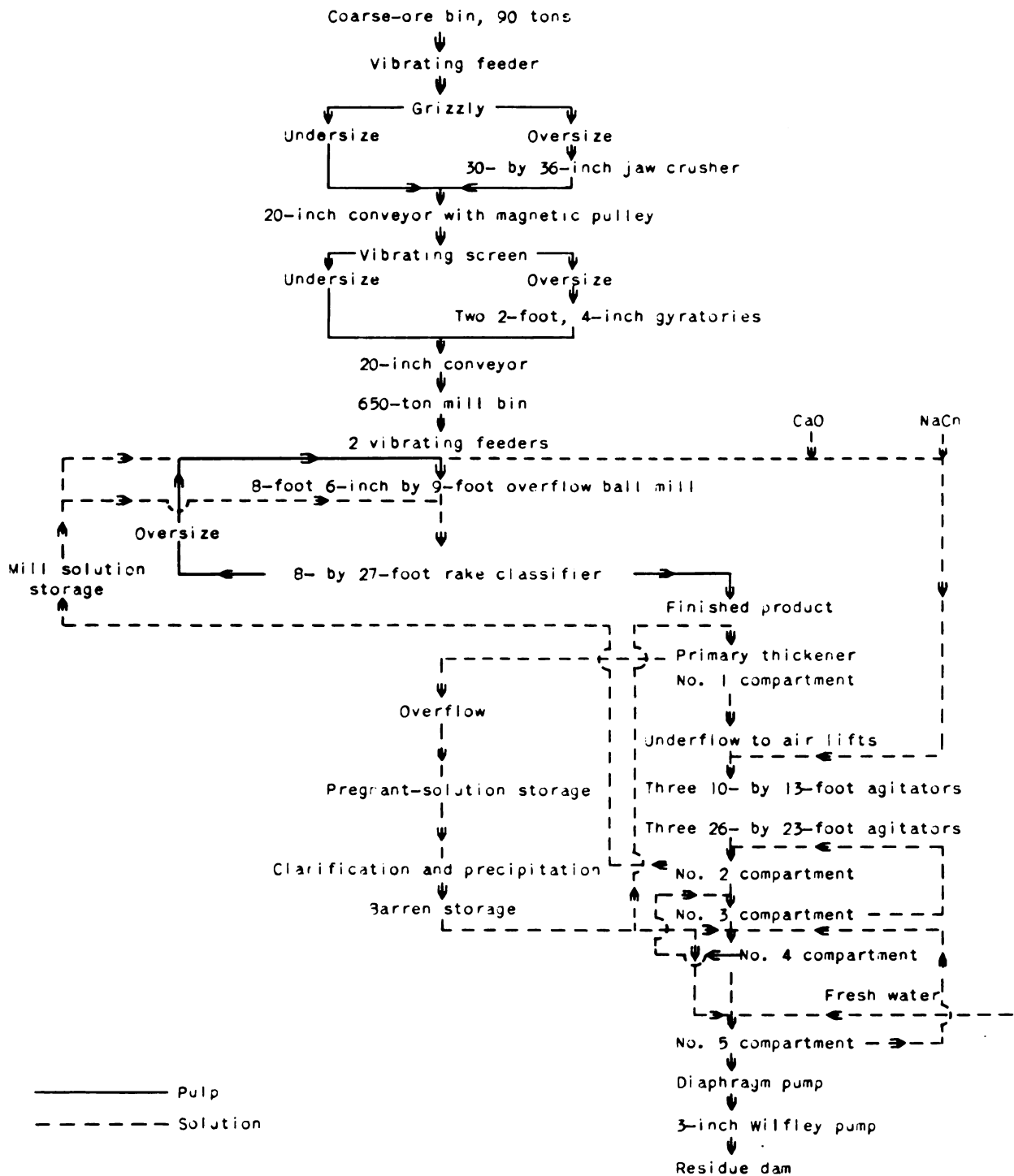
At present the mine is being worked through a vertical shaft 640 feet in depth. This shaft, consisting of a single hoist compartment and a manway, has stations cut at 125-, 300-, 400-, and 600-foot levels. On the 300-foot level is an adit 700 feet in length, which provides a second exit.

A steam hoist with wood-fired boiler delivers the ore by skip to a 100-ton bin on the surface whence it is trucked to the Knob Hill mill. An electrically driven compressor with a capacity of 670 cubic feet of free air per minute replaced an older, smaller unit.

The mine is producing about 75 tons of milling-grade ore daily. The present ore reserves and recent development work are encouraging enough to warrant increasing ore production to 100 tons daily.

Prospecting is now prosecuted by long-hole drilling. The equipment comprises a heavy-duty Leyner drill using 30-inch sections of 1 1/4-inch round drill steel; the steel is coupled, making a unit capable of drilling holes 80 feet long. The sludge samples have proved reliable; this method of prospecting is attractive here because it combines economy and speed with satisfactory sampling results.

As drilling is scattered widely throughout the mine, jack bits are used in regular operations to save expense in transporting steel. The ground which stands well and requires little timbering, is conducive to shrinkage stoping. A portion of the vein near the surface is mined as a glory hole.



Note: All thickener compartments in one tank

Figure 1.- Flow sheet of Knob Hill mill, Republic, Wash.

The mine has suffered in the past from uncontrolled leasing operations, which have left it in poor shape both for mining developed ore and for carrying out an exploration program.

The personnel consists of a crew of about 25 men divided into two shifts and supervised by a mine superintendent and night foreman. All mechanical work is handled by the mine-shop force at Knob Hill. Likewise, all engineering, assaying, office work, and general supervision are handled by the Knob Hill staff.

MILLING

General

The cyanide process for the dissolution and recovery of precious metals is employed at the Knob Hill mill. The ores must be ground relatively fine to liberate the precious metals. A countercurrent-decantation thickener system is used for replacing the enriched solutions. The nominal daily capacity of the plant is 400 tons, although variations from 25 percent below this figure to 45 percent above it have been maintained for weeks at a time; therefore, the mill may now be rated at about 450 tons daily average.

Early tests indicated the necessity for rather fine grinding, and the necessity for housing and heating the mill dictated the application of the so-called all-sliming method of treatment. Although the flow-sheet remains, in substance, similar to that originally contemplated, operating experience has brought about many changes.

Construction began in the autumn and continued throughout the unusually severe winter of 1936-37. Although 3,000 yards of concrete was poured for this mill in midwinter, including one severe cold spell of 3 weeks duration when the temperature seldom rose above zero, nevertheless it was found necessary to condemn less than 3 percent of the concrete.

The mill began operation in June 1937; the following description of the mill is as of October 1939. The Knob Hill mill is simple in design and follows conventional lines, as the accompanying flow sheet will disclose (see fig. 1). At present, grinding is done in a cyanide solution, although originally a water grind was planned to permit neutralization of soluble ferrous salts that, according to a mill test of a sample of development ore, occurred in the ore. However, these salts have not yet been found in appreciable quantities in freshly broken ore. Pulp flows from the grinding circuit to a primary thickener, where some dissolved values are stripped from it. The thickened pulp then passes through the agitators for more complete dissolution of the metals. Replacement is accomplished by a four-wash, countercurrent-decantation system, after which the impoverished pulp is pumped to a residue-storage dam.

The milling cycle can be divided into three phases, which include (1) preparation of the ore, (2) reduction to proper size and condition, and (3) treatment. A description of the equipment used in these steps, together with a discussion of its application, follows.

The mill lay-out is conducive to efficient operation. A main operating floor extends around the grinding section, agitators, and thickener and is either at the same elevation or above the top of all main tanks, making the equipment easily visible. All piping is carried above or below floors, with pumps situated on a floor about 25 feet below the main floor. The precipitation equipment is on the floor level with the pumps, and the furnace room is on the bottom or thickener bottom level. Benches, sinks, glassware, and other control equipment are arranged on the main platform so that the operators can perform their test and sample work without losing sight of the main units.

Innovations in the mill during the past year included the installation of a semiportable, 1-ton, reversible, electric hoist with a rope speed of 30 f.p.m. It is equipped with a magnetic, automatic brake with remote control; it can be utilized anywhere around the mill, being easily transported because of its moderate weight, which is less than 500 pounds. This machine is used primarily for lifting liners in and out of the ball mill, where it has proved invaluable in giving the crew inside direct and complete control when moving these pieces. It is also used to reload the ball charge. A bottom-discharge bucket with a capacity of 1 ton of balls is dragged up a skid on the side of the ball mill to a position over a vertical manhole, wherein a large, heavy metal funnel has been placed. A blow opens the bottom of the bucket, discharging the contents into the mill. One man stationed on top of the drum and three or four men shoveling into the bucket on the floor can reload 30 tons of balls in 1-3/4 hours by this method. Ratcheting socket wrenches with handles 3 feet long also materially speed the tightening of nuts on the 1 1/2-inch liner bolts. These two devices have virtually cut the mill relining time in half.

Crushing

A wood rectangular bin with a live-load capacity of about 90 tons receives the ore as delivered from the mine in dump trucks. The ore forms its own slope and bed. No grizzly is used over this bin; because the ore broke coarser in mining than was anticipated, several changes in construction were required to accommodate large boulders. In the past few months, improved practice in blasting has resulted in more efficient breaking, and fewer large boulders, and has rectified the trouble so that most of the ore now received is suitable for the design of the discharge gates and feeders.

Ore is discharged from an opening in the bottom and front of the bin onto a Jeffrey vibrating feeder 3 1/2 feet wide and 6 feet long. The feeder is operated by a motor-generator set having a range of 0 to 26 amperes. The ore gates originally installed to control the discharge of the ore were found unsuitable and were replaced by a series of steel baffles outside the bin, which absorb the shock and prevent rock from flying beyond the feeder when dumped into an empty bin. A rack-and-pinion gate beyond and below the baffles serves only to regulate the flow of the larger rocks. This gate and the vibrating feeder can be regulated by remote control so that the crusher operator, stationed at a strategic point, can observe the flow of material through the jaw crusher and loosen any ore blocked in the opening of the bin.

Ice and frost in the ore during the winter and spring months presented many problems. As it was difficult to keep the ore moving in the bin when it was full, it was found necessary to dump no more than 10 or 15 tons into the bin at a time. When trucks dumped into the virtually empty bin, impact shocks heavier than those which it was designed to withstand were exerted on the front of the bin. To prevent complete destruction of the front of the bin, a steel curtain is now suspended inside the bin about 5 feet from the front. This curtain is hung from the top on rods, so that it may swing freely. The impact resulting from dumping the trucks is absorbed by the curtain without damage to it or the front of the bin. The curtain does not restrict the flow of ore when the bin is full.

Building up of sticky or frosted fines on the vibrating feeder, which gave much trouble, was virtually eliminated by installing a false bottom in the feeder, which is bolted only on the sides. This not only increases the slope of the feeder bottom but also supplies a vibration independent of the feeder itself although actuated by it.

The vibrating feeder discharges onto a stationary grizzly 28 inches wide by 36 inches long with 4-inch openings. Undersize from the grizzly drops through a vertical chute onto the coarse-ore conveyor belt, while the oversize goes to a 30- by 36-inch, type-H, Traylor jaw crusher driven at a speed of 250 r.p.m. by a 75-horsepower motor through V-belts.

No reliable estimate is available of the proportion of ore that actually passes through the jaw crusher. The nature of the ore, together with the methods used in open-pit mining, tends to yield a product of unpredictable and variable physical character. Such a product precludes an accurate estimate of work done by the various units in the crushing section. Possibly, 20 to 35 percent of the ore actually is broken by the jaw crusher. Using this yardstick, the life of the lower plate on the swinging jaw is estimated to be about 60,000 tons crushed. Owing to the lenticular shape of the boulders, the opening of this crusher can be closed to 3 and sometimes to 2 1/2 inches. The loss of scrap is relatively high, and it has been found economical to hard-surface crusher plates, a procedure that pays good dividends when the facing is applied to new plates and is renewed as soon as worn off. A hard surface is applied also to the stationary lower plate, which will crush about three times as much as the swinging jaw plate before repair or replacement is necessary. The upper plates appear to have an indefinite life.

The discharge from the jaw crusher joins the undersize from the grizzly on a 20-inch conveyor running at 250 feet per minute and chain-driven by a 2-horsepower motor. This belt travels directly under the jaw crusher, and material is fed to it through a poorly designed arrangement that allows the sharp fragments to strike the belt at a right angle to the line of travel, with resultant heavy wear. The present belt is 5-ply construction with a 3/16-inch top rubber cover; it is the second belt that has been applied to the work. Many modifications have been made to soften the cutting action of the rock that falls from the crusher; the estimated life of the present belt is 400,000 tons.

The driver pulley of this conveyor belt is a Dings magnetic pulley 16 inches in diameter, with a 22-inch face, and drawing about 7 amperes at full load. Although this pulley gave satisfactory service at the original belt speed, it no longer offers adequate protection to the secondary crushers at the present rapid belt speed, especially when the belt carries peak loads. A permanent magnet suspended over the belt near the discharge possibly would better insure removal of such "tramp" material as dipper-teeth sockets and other metal lost in the mining process.

The conveyor discharges onto a 3- by 8-foot Allis-Chalmers horizontal, double-decked, vibrating screen driven through V-belts by a 5-horsepower motor. No fixed size of screen openings is employed, as both winter weather and variations in types of ore impose conditions that must be overcome by varying the screens. The upper deck normally is mounted with a screen having 2-inch openings, but the lower-deck screen openings range from 3/8 inch, when extremely hard but dry rock is crushed to 1 1/2 inches when frosted or frozen fines are handled. Undersize from this screen passes down a steel hopper chute onto the main conveyor, while the oversize goes to one or both of two 2-foot 4-inch, type TY, Traylor gyratories driven through V-belts by 60-horsepower motors equipped with line starters.

The eccentric swing of the gyratories is 5/8 inch. Although originally the feed opening was 9 inches, this was reduced to 5 1/2 inches later through installation of special upper plates supplied by the manufacturer in an effort to reduce the excessively high proportion of plus-3/8-inch product. Although this change accomplished the desired result only in part, the large pieces produced showed a noticeable improvement in uniformity of size. The life expectancy of the upper plates is about 3 years. Restriction in feed openings made necessary reduction in the size of maximum pieces delivered to the gyratories. This was the reason for closing the discharge opening of the jaw crusher, as mentioned above.

The gyratories were designed to use "standard," "oversize," and "double-oversize" lower concaves. The theory of this design is to wear out mantles completely without sacrificing crushing efficiency by replacing worn-out concaves with oversize renewals. The economy of this design is fully successful only when the mantle lasts as long as two concaves. Moreover, it has been found that the annular grooves or rings formed in the concaves and mantles when the machines are set for a fine discharge are the source of considerable trouble when a new part is used in conjunction with a partly worn part. A new design, modifying the curve at the lower end of both concaves and mantles, has been evolved by the mill staff and approved by the manufacturer. It is believed that the new design will partly overcome excessive wear. Splitting the mantles horizontally just above the wear line is expected to reduce scrap loss further.

Owing to the variable character of the ore, no satisfactory data are available concerning the percentage of total feed passing through the gyratories. On the basis of 60 percent passing through, a reasonable estimate, the life of manganese-steel lower concaves and of mantles is about 25,000 tons each. Molybdenum-chrome alloy-steel lower concaves are being tried, but have not been in service long enough to warrant estimating their comparative worth.

The eccentric-shaft speed of the gyratories is 250 r.p.m. Discharge openings, measured on the closed side, range from $7/16$ to $3/4$ inch, depending on rock and mill conditions. The original plan called for a continuous maximum-discharge opening of $3/8$ inch; however, it has not been possible to decrease the maximum size of material reaching the gyratories to a size that will permit using this opening under present operating conditions. With the gyratories set at $1/2$ inch and the rock medium hard, both machines must be in service when 60 tons or more per hour is passing through the crushing section. Maximum size of particles passing the jaw crusher will vary according to the breaking characteristics of the ore and may be as large as 8 inches long by 6 inches wide or, if blocky, $4\ 1/2$ inches measured on the longest axis.

The discharge of the gyratories joins the undersize from the screens on the main or mill-bin conveyor. This belt is 20 inches wide and runs up a slope of nearly 21° at 175 feet per minute. It is driven by a $7\ 1/2$ -horsepower gearhead motor connected directly to the head pulley shaft. The belt is of 5-ply construction with a $1/8$ -inch rubber covering; the period of efficient usefulness is expected to exceed 5 years.

All motors in the crushing section are controlled from the operating floor, which surrounds the opening into the jaw crusher at the same level. The building and units follow a "straight-line" layout, so that the operator from this floor may look down through the open bay and see all units. Motors are started through progressively interlocked switches from the main conveyor to the coarse-ore feeder. Stopping any unit automatically stops all units ahead of it. An exception is made for the gyratories, which may be controlled independently in or out of the interlock.

Owing to the necessity for 50-percent overload relays in the switches the gyratories often slow down without immediately stopping. Despite the automatic shut-down switches, so much feed is accumulated in the units during the slowing-down period that costly delays may ensue while a gyratory is cleaned. To overcome this, howlers and lights actuated by governors on the counter shafts have been installed to warn the operator immediately when a machine is retarding. If the operator does nothing about the signal, the governor then stops all units ahead of the gyratory before the motor switch actually opens to stop the gyratory.

Two men operate the crushing section. Although the original design contemplated but one operator, the second man has proved his worth by reducing lost operating time. This, in turn, tends to level off mining operations and actually permits a reduction in mine personnel. All units, except motors and their supplementary parts, are serviced daily by the operating crew under a strictly maintained schedule. Maintenance and repairs are in the hands of the mill repair men, who make all major repairs at night. Under this system, little lost mining time has ever been chargeable to crushing-section delays.

The head crusher operator reports daily, on a suitable form, the number of hours operated, the number of hours each gyratory has run, the setting of each (which is measured daily), time lost and cause, and the number

of truckloads of ore delivered to the coarse-ore bin. This information is transferred to permanent records.

The main conveyor discharges into a circular steel bin 28 feet in diameter by 40 feet in height. Its nominal capacity is 800 tons, but the actual live-load capacity is 625 tons when completely filled. An unsatisfactory tendency toward segregation of fine and coarse particles in this bin was due to the original single center-point discharge. This has been partly remedied by distributing the discharge through rubber-lined steel chutes, thus forming cones over the two feeders in the bottom.

Grinding

The bottom of this circular steel bin is composed of a concrete slab with a tunnel in it that contains two equally separated discharge chutes. The ore drops on two 18- by 40-inch Jeffrey vibrating feeders, each actuated by its own motor-generator set. These two units feed the ore onto a 20-inch conveyor belt running at 40 feet per minute, which in turn discharges into the feed drum of the ball mill. The tunnel is large enough to contain all the above equipment with ample room for servicing it. The mill bin is within the mill building, and the main conveyor is housed in a wood sheathed tunnel. Temperatures are always above freezing in the bin and conveyor tunnel. The vibrating feeders are interlocked with the ball-mill motor control and cannot be started until the ball-mill motor is running; they stop when the ball-mill motor circuit is broken.

The ball mill is a Traylor overflow type with 30° ends; the inside diameter is 8 1/2 feet, and the length of the cylinder is 9 feet. It is equipped with a combination drum and scoop feeder, the drum having a diameter of about 30 inches and a 16-inch opening, while the scoop has a radius of 5 feet 9 inches. The mill is driven at 19 1/2 r.p.m. by a 300-horsepower slip-ring motor. The speed reduction is obtained through a double-reduction gear train mounted in and completely enclosed by a leakproof case. The main gear is mounted on an extension of the discharge trunnion beyond the discharge-trunnion bearing. The gear-train shafts are carried on roller bearings. Gears and bearings are lubricated through a combination splash and pump system from a well in the case, the capacity of which is 110 gallons of oil. A gear pump draws oil from the sump and delivers it to the bearings and contact points of the gears through a water cooler and filter. The main gear runs through the oil reservoir and throws a large volume of oil to all parts of the case. Oil consumption in the case is nominal, amounting to around 5 gallons a month. To date it has been found unnecessary to renew the oil as the temperature has consistently averaged only 20° above room temperature. The feed trunnion is a self-aligning babbitted design and is lubricated by grease. The internal diameter of the trunnions is 20 inches.

During the better part of the operation, the Williamson "tear-drop" or knobby type of liner has been used in the ball mill. The weight of a full set is about 37,000 pounds, and scrap loss is now 15 to 20 percent.

The life of a manganese-steel liner of this type is about 4,000 hours, during which time approximately 70,000 tons can be ground. As the liners of this design have a very thin section at some points, brittle alloy steels do not seem to give satisfactory results.

Several designs of liners have been tried in an attempt to obtain as long life for the end liners as for those in the breast. Originally, the breast liners outlasted two sets of end liners; it has been found that the mere addition of weight to the observed wear points is not completely successful in remedying the wear. Reversible lifters are now being tried, and evidence indicates they will last as long as the breast liners. Economy, however, points to another design, the manufacture of which was tried earlier in one end of the mill; it had a life equal to that of the breast liners. In this design the metal is distributed according to the calculated and observed motion of the balls in the tapered ends. It ignores the wear points shown by removed sections and takes into consideration the logical path of the balls. It is the only design that has achieved this result economically.

Until recently, only 4-inch balls were fed to the mill; the present feed is about two-thirds, by weight, 3-inch balls and the remainder 4-inch balls. The mixed feed may have increased the mill duty by at least 10 to 12 percent; this has been made possible through the improved mixture of ore coming from the mine. Forged, alloy-forged, and cast balls have been given exhaustive tests with no outstanding superiority of performance indicated for any type. Under the circumstances, cost seems to be the controlling factor. Consumption has ranged from a low of 2.3 pounds per ton of ore ground to a high of 7 pounds over periods of several weeks, irrespective of the type of ball. So far it has been observed that the character of ore being ground directly and chiefly influences the consumption of grinding media steel. The consumption of balls during the life of the operation is approximately 4.5 pounds per ton of ore ground.

The mill is equipped with an indicating watt-hour meter, which not only supplies information as to power input but also controls the feed of balls to the mill. With a normal ore, the highest efficiency is obtained with a power input of about 225 kilowatt-hours, beyond which but little gain in capacity is observed. At this point, with the average type of ore, the mill will contain a ball load of some 30 tons with a circulating load of pulp of about 500 percent. When soft, porphyritic, foot-wall ore is handled, the input of power is permitted to drop to 185 to 190 kilowatt-hours, at which point the circulating load is virtually nil.

The ball mill is in closed circuit with an 8- by 27-foot Model DSFX Dorr classifier running at 17 strokes per minute and driven through V-belts by a 15-horsepower gear-head motor. The specific gravity of the overflow is varied according to the feed and ranges from 18 to 23 percent solids. The circulating load depends entirely on the ore. Broadly speaking, when ore of normal hardness is being ground, the feed is regulated entirely by the circulating load, which is kept at a certain depth with relation to the classifier rakes. This seems to result in obtaining the maximum tonnage

with this type of feed; moreover, the thickener then has a capacity equal to or greater than the mill. With soft ores the mill has a capacity greater than that of the thickener. To help equalize this unbalanced condition, coarse feed is sent to the mill, the ball load is permitted to drop, the overflow is thinned out, and every effort is made to keep the mill in continuous operation without excessive consumption of steel. Routine practice also includes varying the specific gravity of the ball-mill discharge according to the feed, with a range of 65 to 75 percent solids. One controlling factor is maintenance of a pulp viscosity that will prevent undue proportions of uncracked particles of ore from floating out of the mill. A mechanical brake for the classifier was installed by the mill crew to permit easy and positive control when the heavy rakes are lowered. This device is now a part of the new machines.

One operator a shift controls the ball mill. Balls are fed each shift in accordance with instructions from the foreman. A steel channel section leads from the ball-storage bin to a suitable down tube, which terminates in the drum feeder. Balls are raked into the channel and roll by gravity down to the spout and into the feeder. This permits accurate count of the balls fed and makes possible rapid charging of several tons of balls. Every tenth ball is retained, and the entire sample is weighed; the feed for a 24-hour run is calculated from this weight. The ball-mill operator records on a daily report the balls fed, watt-hour meter readings twice a shift, time when feed or mill is down and the reason therefor, hourly determination of specific gravity and the tonnage being handled by the primary-thickener-discharge pumps. In addition, he calibrates and records the ball-mill discharge and classifier overflow every 2 hours. This information, or such of it as is deemed necessary for permanent records, is consolidated in the metallurgical control books. The specific gravity of the pulps in the grinding circuits is set by the foreman according to conditions that can be anticipated immediately. Enough leeway is allowed so that in an emergency the ball-mill operator may exercise his judgment.

Treatment

The overflow from the classifier flows by gravity into the primary or top tray of a five-compartment Dorr washing thickener 60 feet in diameter. This machine consists of four washing compartments all having an effective depth of 7 1/2 feet, and one much shallower primary tray sealed from the lower compartments by an air lock. The total height of the tank is 39 feet.

This type of thickener has one central shaft upon which all rake mechanisms are suspended and which is driven by one gear. It consolidates all thickening and washing operations in one tank, the various steps or phases being carried on in compartments imposed vertically, one upon the other. The main gear of the machine at Knob Hill is driven by two pinions mounted on opposite sides of the gear and actuated by two 1 1/2-horsepower gear-head motors through chain drives. Care is taken to use motors of the same make and of exactly the same characteristics, so that the speeds will be synchronized. Circuits are so arranged that while separate heater elements

and relays protect each motor against electrical overloads, stopping of one motor will stop the other; both are started simultaneously by the same switch. Each pinion is provided with protection against overload of the gear, shaft, or rakes through sliding blocks, which are actuated by the thrust on the worm. At a predetermined thrust pressure an alarm will sound; if the trouble is not corrected and the pressure is allowed to build up to a greater predetermined point, the motors will be stopped through a circuit that stops both motors regardless of the one affected. Tension on the thrust-absorbing springs of each driving unit is equalized with respect to the other as nearly as possible and can be adjusted to a point where the indicating arms, outside the thrust boxes, will show the actual situation during operation. The mill staff improved on the alarm device supplied by having the first stage of excess load pressure sound a horn; then, when the pressure increases enough to stop the machine, the horn ceases and a bell rings until the motors are started again. This gives the operators full information regarding any abnormal condition in the thickener without requiring them to watch either the machine or the indicating lights. Rakes are lifted and lowered by means of a planetary gear-reduction unit mounted over the shaft. Even with a ratio of 200 turns to an inch of lift, the rakes can be raised slowly by two men only with great effort. Eighteen-inch extension handles attached to the lifting wheel materially assist in rapid lifting of the rakes at night, when only two men are available for this arduous job.

A No. 4 Inspiration-type Dorrco triplex diaphragm pump and a 4-inch discharge-column air lift using air at about 12 pounds effective pressure are employed to elevate primary-tray thickened pulp about 6 feet above the thickener top to the first of the agitators. The Dorrco pumps came as original equipment but, for reasons to be described later, were found to be of adequate capacity. After several other methods were tried, the air lift was finally inserted permanently into the circuit as best suited to the job. Present practice is to use the diaphragm pumps only when the volume of pulp exceeds the capacity of the air lift.

The agitation system consists of three tanks, each 10 feet in diameter by 13 feet in depth, and three tanks, each 26 feet in diameter by 23 feet in depth, all equipped with Dorr type A low-head agitators. The small machines revolve at 6 r.p.m. and are driven by 1 1/2-horsepower gearhead motors through V-belts, while the larger units turn at 4 r.p.m. and are driven in the same manner by 3-horsepower motors.

Within 3 months after the mill went into operation, it was discovered that the pulp was building inward from the sides of the agitator tanks and outward from the central column. After various devices failed to control or stop this, 1-inch pipes were suspended from the end of one distributor arm of each agitator about 15 feet down the side of the tank. Air was introduced into this pipe, which not only successfully controlled the growth from the tank sides but also gave additional aeration. To bridge the gap between the lower end of these pipes and the rake arms, a pipe about 6 feet long was attached vertically to the end of one rake arm.

The design of the agitators did not permit introducing air into these 6-foot pipes without completely disassembling the mechanism and rearranging the piping system, a task hardly warranted by any possible improvement over the method used. Air was introduced into another pipe attached to the central column and extended into the pulp, and further growth was stopped at this point.

No accurate estimate of the volume of air normally used for agitation is available. The compressor employed is rated at 250 cubic feet of free air per minute at 25 pounds and is driven by a 25-horsepower motor through V-belts. Ordinarily, the pump is working about three-fourths of the time, which may indicate to some extent the amount of air normally used.

The pulp flows by gravity from one agitator tank to another. A system of by-passes was installed during the first year's operation, so that at present stopping of any or all agitators does not necessarily require shutting down of the mill, but merely involves a higher loss of residue while the unit (or units) is being put into operation again.

From the last agitator the pulp flows into the first of the washing trays of the thickener, which is the second compartment. From thence it travels downward through the four lower compartments, while the solution rises countercurrent to its flow in accordance with the differential specific gravity balances upon which the tray thickener depends for functioning. The thickened pulp or residue is removed from the bottom compartment by a No. 4 Dorco triplex diaphragm pump powered by a 5-horsepower gear-head motor direct-connected to the eccentric shaft. The residue pumps are bracket-mounted on the side of the thickener tank, with the discharge lip about 1 foot above the top of the tank. They discharge into the sump of a 3-inch, high-head, Wilfley pump, which is also mounted on the side of the thickener tank about 30 feet above the bottom of the tank. The Wilfley pump is connected directly to a 40-horsepower motor running at 1,750 r.p.m. A maximum pumping pressure of 100 pounds per square foot is developed by the pump when 14-inch impellers are used in delivering residues to the storage dam.

Impellers in this pump last about 15 days. When new casings are used, they have a life of about 30 days. It has been found, however, that worn casings can be repaired with chilled cast-iron welding rod, using the arc. This process can be repeated until the metal becomes too thin to withstand the pressure head; the life of a casing thus can be virtually tripled. Rubber-faced follower plates last 4 to 6 months.

Residue pulp is delivered to the residue dam through a 4-inch o.d. light-steel pipe line. No appreciable wear has been noted in this line as yet. The nearest point of the dam is about 700 feet from the mill. The crest is 1,000 feet in length on the long side. The dam is built in the conventional manner by discharging pulp along the crest inside face boards, through hoses attachable to nipples in the main pumping line. To facilitate movement of the main line inward as the dam raises it, the pipe is broken into 20- and 40-foot lengths connected by short pieces of hose fastened to

the pipes with heavy-duty clamps. Owing to the fine grind, the nature of the slimes, and other operating conditions, trouble has been experienced occasionally in separating the slimes from the sands enough to permit rapid and complete draining of the sands. Discharging the hoses against a rubber-covered plate tends to separate the water and slimes from the sands and helps to pack the sand firmly against the face boards. However, to safeguard the dam further, the face is raised at a slope of $1/2$ to 1 instead of the normal 1 to 1.

Winter conditions at Republic require special treatment in storing residues. Owing to the relatively small area within the dam, it was impossible, during the first winter, to avoid placing most of the residues on the face of the dam. Naturally, these froze before draining, and during the spring thaw only the heroic action and resourcefulness of the entire mill crew averted a major catastrophe. During the colder months of the second winter there was area enough inside the dam for the residue, and little difficulty was experienced when the thaws came. It is believed that all serious trouble from this source has been eliminated for the future.

The Merrill-Crowe simultaneous clarification and precipitation process is used to recover dissolved metals. The clarification equipment consists of 20 leaves in one round tank and 8 leaves mounted in a rectangular tank. All leaves are 8 feet long by 5 feet wide and are covered with muslin without burlap undercovers. The manifolds of each tank are connected to the main pipe leading into the deaerating tower. The life of leaves is about 3 months in the main tank and four to six times that period in the auxiliary tank. Covers are not rotted when removed but have become too blinded for further use. Salvaging used leaves is uneconomical. Replacement of the entire set of 20 leaves is made at one time and requires about $1\frac{1}{2}$ hour's work of three men. Leaves are not removed from the tank for washing, and all are washed at the same time. In fact, a leaf never is removed from the tank from the time a new cover is put on until it comes off; as a result, no trouble has been experienced with holes in the covers, and bag pressures caused by solids in the solution going to precipitation are unknown at Knob Hill.

It is necessary to wash the large unit about every 5 days. This takes about 2 hours actual time, but, since this operation is handled by the solution operator along with his other duties, the elapsed time may be longer. For this reason, the auxiliary unit of eight leaves was installed, making it unnecessary to reduce the flow of solution going to precipitation for more than a few minutes at a time. These leaves can clarify the entire flow of solution for several hours, which is an important advantage in many ways, especially if solution storage capacity is limited.

Diatomaceous earth is used as an aid to clarification. When new covers are installed, crystal-clear barren solution is used to fill the tank. Hy-Flo, which is made up in a small tank mounted over the clarifier unit, is then added as a slurry, 4 to 5 pounds being used per 100 square feet of filtering area. Precipitation is then stopped and the discharge from the bag pump is diverted into the clarifier tank through a maltese-

cross aperture in the bottom of the tank; this creates a rising current and tends to keep the Hy-Flo in suspension. Although a special pump probably would be more efficient in obtaining large-volume circulation and avoiding precipitation shut-downs, nevertheless, large-volume circulation is obtained and all leaves are quickly and uniformly coated with Hy-Flo after which the pump discharge is returned to its normal flow through the bags. In this way the first solids to reach a new cover are the particles of Hy-Flo. The same handling cycle is followed when leaves are washed. This method prevents any mud that may have been driven into the cloth from getting into the precipitation bags, and an uncontaminated coating of Hy-Flo is assured.

Provision is made for continuously feeding Hy-Flo to the solution going into the clarification tank by means of a standard belt feeder somewhat larger than the customary zinc-dust feeder. However, continuous addition of Hy-Flo has been tried, but its economic advantage has not been demonstrated at Knob Hill when the clarifying capacity of the leaves is considered.

Zinc-feeding equipment comprises the standard Merrill belt feeder and mixing cone. A mercooid switch actuated by the float lever arm of the de-aerating tower sounds a howler should the solution level in the tower drop below a predetermined height. When zinc is added to the feeder belt, a block is placed on the belt in a position to touch the howler 15 minutes before the last of the zinc is fed off the belt. Mercooid switches also give warning should the barren-solution pump become air-locked and allow the bag tanks to overflow. All precautions are taken to insure continuous operation of the precipitation system; for this reason, the warning system is considered better than a method in which the flow would be stopped during emergencies.

The precipitate-collecting system consists of three tanks with 64 bag headers in each. Only two tanks are used at one time, and these do not always contain a full complement of bags. Bag heads are of a new design that clamps the bags to the head instead of requiring the old elementary tying.

The new head offers many advantages over the older ones. No experience or tools are required to attach bags securely to the heads, as finger pressure on the wing clamping nut is adequate to provide a nonleaking grip greater than the bursting strength of the cloth. Moreover, tying bags was a tedious and slow job exceedingly hard on the fingers when a large number of bags was to be prepared. Bags often are clamped on the new heads at the rate of 100 per hour, and bags can be removed virtually as quickly and easily. One-half to two-thirds of the usual time required for removing and replacing bags from a 125-bag unit can be saved; moreover, life of the bag also is increased.

Ten days elapse between clean-ups. On the morning of a clean-up, the solution operator starts one unit draining, leaving only one in operation. As soon as the bags have drained, which is usually about half an hour, they are quickly removed by the clean-up man and fresh bags are screwed on. The

second unit is then shut down to drain. The third or stand-by unit, with its complement of bags, and the first unit are then ready for operation. From 8 to 12 pounds of Hy-Flo mixed into a slurry is added through the mixing cone. When the walls of the bags have had time to become firmly coated with the mixture, 10 to 20 pounds of zinc dust is added manually as quickly as it can be passed through the mixing cone. During this period, and for 45 minutes thereafter, discharge from the bag tank is circulated through the clarifier tank. The bag tail is sampled at the end of the 45 minutes, and the normal feeding of zinc is begun. It might be added, however, that it has been found profitable to increase the zinc-dust feed about 30 percent above normal for the first 24 hours after a clean-up, decreasing the excess to 15 percent for the second day and dropping to normal during the third. Four to five pounds of lead acetate is added to the mixing cone with the initial charge of zinc. Thereafter, it is fed in drops to the clarifier tank at the rate of about 1 1/2 pounds in 24 hours.

As the bags are removed, they are dumped into a 5- by 3-foot redwood tank, brushed lightly, and piled for washing. When all precipitates have been collected, the contents of the tank are discharged onto a wooden filtering tank 7 feet in diameter for removal of excess moisture, after which they are transferred to pans and dried overnight in an electric drier. The solution operator assists the clean-up man for about an hour in the morning to start this work after which the clean-up man works alone and usually finishes this phase of the clean-up in 5 or 6 hours.

Melting is done in a No. 275 Monarch tilting furnace fired with Diesel fuel oil. Five hundred to seven hundred pounds of precipitate is melted at one clean-up, 12 to 15 hours being required to complete the process. A carborundum-type crucible is used in preference to the clay-lined graphite models. The controlling factors in melting are life of crucibles and fineness of the bars. Purity of the bullion improves materially when clay-lined crucibles and an oxidizing flux are used, but the saving in minting charges does not offset the additional melting cost owing to the shorter life of the clay-lined crucibles. Total fineness ranges from 825 to 850.

The charge consists of about 50 parts borax glass, 30 parts soda, 30 parts silica, and 3 parts fluorspar; and 100 parts of precipitates. Selenium is noted in the gases expelled during melting. At prevailing prices, slags average about 50 cents a pound in total gold and silver content and are sold to the smelters. The slag of each melt is sampled by stream-cutting the pour of slag and granulating the sample in water. The charge is added to the crucibles until the molten slag is near the top when the slag is poured off to a point near the metal line and charging is resumed. This procedure continues until enough metal is accumulated to be cast in full-size bars. The bars are sampled by drilling. Prolonged comparative tests of drilling and streamcutting the metal during the pour failed to disclose any material difference, although if large enough quantities of metal were being handled, stream-cutting might produce a more accurate result. The clean-up man alone handles the melting, except when casting bars, when an operator is present to guard against accidents.

After the melt, the clean-up man puts the refinery in order, sacks slag and washes bags in a household-type electric washing machine. Oxydol soap and hot water have been found to be an effective combination for keeping bags clean and soft. The partly dried bags are clamped to the spare heads and stored. The whole operation, including melting and all other regular work, from one clean-up until the entire set of bags is made ready for the next clean-up, requires about 3 1/2 days of the clean-up man's time, after which he is free for other duties.

Mill Operation

Metallurgical Control

The mill is operated to return a maximum profit to the company. This involves balancing ultimate recovery against maximum daily tonnage, taking into consideration costs of all kinds., grade of ore with its physical and chemical milling characteristics, and life expectancy of the mine. As the grade and character of mill feed vary considerably, the manipulation of all units influencing the result is not fixed but varies from time to time to meet, as nearly as possible, the requirements of the operating policy. This justifies the rather indefinite limits of mill control indicated by some of the data.

The mill was designed originally to treat ore with the substantial quantity of ferrous sulfate indicated by the composite sample tested. This required grinding in water with preliminary aeration before the cyanide is added and explains the air-locking of the primary tray of the thickener from the lower compartments, as well as the three small Dorr agitators, which were to do the aerating. Milling was hardly under way before it became evident that little soluble ferrous salts in harmful proportions would be found in freshly broken faces, and the plant was converted to grind in mill solutions; in consequence, ores and solutions had more time in contact, making possible higher tonnage without sacrificing recovery.

Grinding in cyanide also contributed to making a coarser grind possible. Originally it was proposed to grind the ore to a point where only 3 percent was plus 100 mesh, with at least 72 percent minus 200 mesh. It was early indicated that this grind would limit the mill to an optimum of 400 tons daily, and that this would not be possible most of the time. Moreover, it soon appeared evident that 90-percent recovery of gold might be anticipated with a coarser grind in solution, and only about 3 percent additional recovery of gold might be obtained by the finer grinding. Operating experience soon indicated that no fixed screen analyses should be demanded of the mill but that the grinding and classification should be governed entirely by the character of the ore. This has come to mean a range of 5 to 12 percent remaining on 65 mesh and 52 to 45 percent passing through 200 mesh. The following tabulations show the percentage by weight of products at various points in the grinder circuit. These analyses cover a period during which most variations occurred in the ore and probably is a fair average of yearly performance.

BALL-MILL FEED

<u>Mesh</u>	<u>Percent by weight</u>	<u>Mesh</u>	<u>Percent by weight</u>
+ 2	37.3	+ 8	11.0
+ 3	13.1	- 8	31.5
+ 4	7.1		100.0

BALL-MILL DISCHARGE

<u>Mesh</u>	<u>Percent by weight</u>	<u>Mesh</u>	<u>Percent by weight</u>
+ 2	1.3	+48	17.8
+ 3	.8	+ 65	13.4
+ 4	.6	+100	7.7
+ 8	1.3	+150	6.1
+10	.7	+200	5.2
+20	4.3	-200	24.3
+35	16.5		100.0

CLASSIFIER RAKE DISCHARGE

<u>Mesh</u>	<u>Percent by weight</u>	<u>Mesh</u>	<u>Percent by weight</u>
+ 2	1.7	+ 48	26.8
+ 3	.9	+ 65	12.5
+ 4	.7	+100	5.4
+ 8	2.0	+150	3.0
+10	1.0	+200	1.7
+20	7.5	-200	8.1
+35	28.7		100.0

CLASSIFIER OVERFLOW

<u>Mesh</u>	<u>Percent by weight</u>	<u>Mesh</u>	<u>Percent by weight</u>
+ 65	10.5	+200	8.9
+100	12.6	-200	56.0
+150	12.0		100.0

Cyanide solutions are kept at a minimum strength of 0.025 percent NaCN during the treatment period; this is much lower than was originally anticipated, but no adverse influence on recovery has been noted. Incidentally, the ores mined from the two company properties are the only ones in the district not benefitted materially by solutions containing 0.05 to 0.10 percent NaCN. An extended series of tests recently made on ores of the other developed mines in the district indicates that unless grinding is carried to nearly 90 percent minus 200 mesh, satisfactory recoveries can be obtained only by applying a strong cyanide solution and then only in conjunction with a

"promotional" salt. These salts include soluble lead salts and mercuric chloride, with economical practice favoring the lead. Such small quantities as 0.1 to 0.5 pound per ton of ore of any of these reagents produce striking results. Mud Lake ore, however, is not benefitted metallurgically by these salts. Aero brand or "black" cyanide is added to both ball mill and No. 1 agitator as a strong solution. A 24-hour supply is made up in one of two small tanks equipped with stirrers. Air is blown into the tank being filled until a test with sodium nitroprusside indicates complete oxidation of all sulfides. A steady-head tank distributes this strong solution to the points of delivery. Daily quantities used and proportions delivered to various points are decided by the foreman.

Lime is added as a milk to the ball mill. A hopper feeds 1/4-inch lime to a flight screw which delivers it into a larger double screw whose blades are partly cut away and turn inward toward each other. Enough solution is added to produce efficient slaking of the material as it travels to the discharge end, where it drops into the sump of a small centrifugal pump. Control of the rate of lime feed is through a U. S. Electric Varia-Speed motor that drives the feed screw. An indicating pointer on the motor showing the final drive-shaft speed is also calibrated to show the rates of feed per shift. Upon the foreman's orders the rate of feed can be changed by merely turning a small hand wheel.

Pulps pass through the agitator system at a density ranging from 30 to 45 percent solids, as determined in the preliminary thickener. The agitators originally were designed to provide 24 hours' agitation daily of 400 tons of solids diluted with 800 tons of solution. No disadvantages, but many advantages with respect to gold, result from carrying a much denser pulp, and no attempt is made to hold down agitator density. Varying the pulp density is one of the control points in obtaining a maximum dollar return.

The mill feed consists of ores with markedly different thickening characteristics. If the settled pulp in the primary thickener begins to rise, the volume discharged will be increased until:

1. The limit of storage in the washing trays and the capacity of the residue pumps at the lowest tolerated density are reached, or,
2. The limit of the water supply is reached, or,
3. A density of the discharge pulp is reached below which no additional solids can be delivered on account of the limiting volumetric capacity of the pumps. This point is found to be about 30 percent solids.

With important exceptions, the same principle governs the control of the residue pumps. Experience has shown that stacking residues of densities below 35 percent solids becomes difficult. Moreover, if residue pumps are run at full capacity with specific gravities below 35 percent solids, the capacity of the water-pumping system is exceeded. Therefore, the lower limit for profitably drawing waste pulp from the thickener is 33 percent solids; this ratio controls the operation of the residue pumps which are operated at higher density only to synchronize with the primary pumps.

Operators are instructed on these points and are provided with charts to guide them when decisions must be made to meet various situations, thus tending at all times to keep the mill operating to the limit of any temporary bottleneck.

This method of control requires close attention to the tray thickener. The revised type of sealing boot between trays installed at Knob Hill was designed to pass 400 tons of solids daily, in the form of pulps with 50 percent solids, when the rakes were in the normal position. Such a density has never been attainable in practice except for short periods and is beyond the thickening capacity of the machine at the greater tonnages commonly handled. As a result, the opening in the boot must be varied from time to time to permit passage of the pulp being handled, but at the same time it must be constricted enough to reduce the tendency of the rising solution to "short circuit." This regulation is obtained by raising and lowering the rakes. A scale, marked in inches, has been attached to the lifting bracket so that a pointer on the shaft indicates the position of the rakes with respect to the bottom of the trays. At each adjustment a record is made of the height of the rakes above datum.

Although the original flow sheet contemplated a washing ratio of 3 parts of water to 1 of solids, the lower specific gravity and increased tonnage required more wash water, so that at present, about 1,600 tons of solution is precipitated daily. The system of piping permits distribution of barren solution to almost any unit in the mill, including any compartment of the thickener. Many combinations of the manner of applying barren washes have been tried without apparent advantage for any particular method. The most satisfactory flow sheet seems to include the addition of all wash water to the bottom compartment of the thickener and the distribution of the barren solutions between this and No. 4 compartment to the capacity of the tray overflow pipes. The rest of the barren solution is added to the primary tray.

Solution Control

As stated before, during agitation the solutions contain a minimum of 0.025 percent NaCN. An alkalinity of 0.02 percent, as determined through titration with phenol, is normal during agitation. The strength of solutions going to precipitation is somewhat below these figures. A small amount of litharge, ranging from 12 to 36 pounds daily, is added to the ball mill, and 1 1/2 pounds of lead acetate is added to the precipitation circuit. The operators titrate and record the cyanide and lime values of the primary thickener, Nos. 1 and 3 agitators, and the overflow and underflow of the bottom tray; but the quantities of reagents to be used are established by the foreman or superintendent, who studies the current titrations and decides the course for the following day.

Since a substantial proportion of the gold is dissolved in the grinding circuit, all primary solutions go directly to precipitation; the remainder of the solution treated at the precipitation unit comes from the

No. 2, or first, washing tray. This is the present normal flow sheet. The piping of the mill, however, has been rearranged and supplemented from time to time until it is now possible to make virtually any change in flow or direction of solutions merely by setting the proper valves; many variations have been tried to determine the effect of the change on the losses of dissolved gold.

Sampling

Sampling for mill control and data is not elaborate. An electrically operated automatic stream sampler cuts about 1 ton from every 400 discharged by the main conveyor. This sample is quartered to 75 pounds, dried, crushed, cut on a Jones splitter, and ground to 100 mesh. A sample is stream cut manually from the ball-mill feed belt at regular intervals and consolidated into one daily sample handled as is the automatic sample. The mill-feed sample, which is used to establish the gross value delivered, represents the ore actually in transit through the mill; the automatic sample is used more as a guide for mining operations, because it supplies information much more quickly and accurately respecting the grade of ore actually delivered during any particular day.

Manual samples are taken hourly from the pump discharge of the primary thickener, the last agitator discharge, and the final residue-pump discharge. The first two are collected daily in strong potassium permanganate solution, cut down to laboratory size, filtered, and washed, and the solids assayed for undissolved metals. The final residue sample is cut and dried without filtering or washing. In addition, a sample of the final residue is taken every 2 hours and immediately filtered, washed, and dried; both pulp and filtrate are saved. Every 24 hours a weighted composite is made from the filtrates and washed pulps and assayed to determine losses of metals. This somewhat elaborate method has been adapted in an effort to determine the real rather than the apparent undissolved and dissolved losses, experience having indicated the impossibility of completely arresting dissolution of values, even in the presence of a strong potassium permanganate solution. This experience is supported by a series of laboratory experiments with gold leaf.

Solutions assayed include bag heads and tails and primary and No. 2 tray-overflow solutions. No less than 17 1/2 assay tons of solution are taken for assay by a modified Chiddy method. Grab samples are taken in an effort to reconcile calculated and actual losses of dissolved values; a long series of special samples, including the samples of the overflows of all trays of the thickener over an 8-months period, was taken for the same purpose. All assays are for both gold and silver.

Mill tonnages are determined by the time-honored specific gravity-time constant volume method. The primary-thickener discharge is calibrated hourly; the final residue-compartment discharge also is calibrated for guidance in operating the thickener and for general control information. This permits operators to control closely the flow entering and discharging

from the thickener and to maintain a balance, or to anticipate and correct an unbalanced condition. This is a very important matter in the operation of the Knob Hill mill, considering the variable-ore feed, thickener characteristics, and operating policies.

Water Supply

The main supply of water for all purposes comes from the San Poil River, on the bank of which the company maintains a pumping station about 3 1/2 miles from the mill. A 9-stage centrifugal pump directly connected to a 100-horsepower motor has a capacity of 130 gallons per minute against a head of 1,000 feet. It delivers water through a 5-inch, o. d., welded-steel pipe line buried 30 inches underground. This pump, together with two horizontal triplex pumps now used as standbys and the pump sump, are housed in a well-insulated house. Strip heaters are employed in extremely cold weather to prevent freezing. Water is delivered to a tank of 80,000 gallons capacity situated at an elevation that will impose a pressure of 75 pounds a square inch at the various operating units in the mill. The original pumping equipment consisted of the triplex pumps now maintained as stand-bys. High attendance and maintenance costs led to installation of a centrifugal pump that could be operated automatically and required only occasional inspection. Operation of the centrifugal pump by a water-level control in the tank required a prohibitive outlay for wiring, so an electric timing clock was installed to govern the pump. This clock actuates the opening and closing of the automatic line-starter switch, which is of the compensating type and tends to reduce heavy surges on the line when such a large motor is started. The starter may be operated manually, with the automatic compensating characteristics retained as an integral part of the starter. The timing clock is adjustable, so that both the on and off intervals can be varied through a wide range; the adjustments have been changed from time to time to meet mill requirements for water. The relatively short interval when the pump is idle was adopted primarily because the water might freeze if the pump was permitted to stand motionless in the line for very long periods during extremely cold weather.

Thermostats are installed for protective purposes on all motor and pump bearings, as well as on the pump case. Overheating of bearings or the pump case owing to loss of primer will shut down the pump. A float switch in the sump will stop the pump before the suction is broken, should the water supply fail. Each of these safety features is connected to a lamp, which glows when its thermostat is functioning. A glance at the instrument, upon entering the pump house, will discover a clue to the cause of stoppage. The pump is inspected weekly by the mill maintenance crew. The whole lay-out has functioned successfully with minimum attention through two winters.

To give complete assurance that trouble at the pump station will receive prompt attention, an indicating lamp in the mill tells when water is being discharged into the tank. The operator, knowing when the pump should be running and when it should be idle, is apprised within a few minutes of any unscheduled stop and immediately communicates this to the foreman.

Solutions which are reclaimed from the residue dam and which supplement the main water supply vary with the season, the tonnage of ore handled, and the percentage of moisture in the pulp; at present these amount to about 20 g.p.m. This water contains about 0.01 percent NaCN and possibly as much as \$0.03 in gold. Additional water is pumped from a well below the mill at the rate of 40 g.p.m.; however, it is fit for use in the thickener only.

Heating of Mill Building

All operating units of the mill, including the fine-ore bin, ball mill, thickener, agitators, precipitation, clean-up, melting, and pumping equipment, together with a 60- by 40-foot tank on one level and a 28- by 40-foot tank on another, are in one large building. The building has a wooden frame, with corrugated iron outside and 1-inch plank sheathing inside. The crusher building is connected to the mill building by a sheathed conveyor tunnel. Owing to its design, it can be sealed only partly against the weather.

Four apartment-house, automatic, oil-fired heaters were bought originally for heating the mill building. It was found possible to mount only three, all of which are on the lower floors. The fourth remained un-assembled during the first winter and was installed later in the crusher building. These heaters were found to be more than adequate, despite the fact that the conveyor tunnel created a strong draft and robbed the mill of considerable heat, which benefitted the crusher building but little. The installation of a draft door has halted this waste. The heaters burn 27° Diesel fuel oil. Air temperature may be controlled thermostatically although this is not deemed practicable in the mill; fans and fuel are controlled automatically. The heaviest monthly consumption of oil for heating the crushing and mill building has been less than 2,000 gallons. The total quantity of fuel for heating the mill buildings for the past winter season was about 6,000 gallons, which, at \$0.09 per gallon, is a reasonable cost for heat.

Daily records are kept of inside and outside temperatures. With the ball mill running, two furnaces have no difficulty in keeping the operating-floor air temperature at about 50° F., although the bottom floor next to the outside walls may be just above freezing. Incidentally, pulp-temperature records also have been kept during winter; these disclose the ball-mill discharge pulp to have a temperature ranging from 68° to 73° F., and solutions to be of a temperature of about 46° F. with a feed full of frosted material. Temperature of the water feed to the bottom tray of the thickener during these periods remains constant at 34° F. for weeks at a time, while the residue pulp leaving the thickener has a temperature of 40° to 44° F., depending on the amount of water being used. This shows a drop of 28° to 30° F. from ball mill to residue pump.

During one period of rather cold weather, when the ball mill was idle long enough for the building to feel the absence of its operating heat, the operating-floor air temperature dropped from 50° to 40° F. As the same number of furnaces was going, this indicates the amount of heat generated by the operating unit.

Personnel

The mill crew, which normally consists of 13 men, including the mill superintendent and foreman, is distributed as follows:

Supervision	2	Clean-up and utility	1
Crushing	2	Maintenance	1
Ball-mill operators	3	Residue disposal	<u>1</u>
Solution operators	3		13

The solution operator is the leader of his shift and is held responsible for emergency decisions covering situations outside the ordinary routine; but he has no control over actual operation of the ball-mill circuit beyond limiting the rate of grind to suit thickener conditions. All mill maintenance and repair, except electrical, is handled by the mill crew. Assaying for mines and mill is consolidated in one office, as are the engineering and local accounting. The office staff normally consists of four persons, including the general superintendent and assayer.

CONSUMPTION OF MAJOR SUPPLIES

The following tabulation gives the consumption of various major supplies in the process of crushing, grinding, and treating 1 ton of ore:

	<u>Unit</u>	<u>Per ton milled</u>
Cast grinding balls	Pounds	3,620
Ball-mill liners	do.	439
Gyratory liners and mantles	do.	036
Cyanide	do.	610
Lime	do.	5,550
Zinc dust	do.	1/ .070
Water consumed	Tons	1.21
Power:		
Crushing	Kw.-hr.	1.09
Grinding, including classification and solution pumping	do.	9.85
Treatment	do.	2.75
Residue disposal	do.	.61
Water pumping	do.	<u>1.64</u>
Total power consumed	do.	15.94

1/Per ton of solution sent to precipitation.

COSTSMining

Mining costs at the Knob Hill mine are based upon the tonnage of ore milled and are shown below with the cost per ton of ore delivered into the mill bin.

	<u>Per ton milled</u>	<u>Total per ton milled</u>
Breaking ore:		
Labor	\$ 0.125	
Supplies	.092	
Power	.017	
Shovel and tractor expense ...	.068	
Total, breaking ore	.302	\$ 0.302
Trucking ore to mill		.079
Stripping:		
Carry-all (\$0.066 per yard)		
Shovel and trucks (\$0.18 per yard)		.291
Total		.672

Milling

Following are itemized the actual and total per-ton costs of milling 16,706 tons of ore:

	<u>Per ton milled</u>	<u>Total per ton milled</u>
Crushing:		
Labor	\$0.031	
Repairs and renewals	.023	
Power	.012	
Total		\$0.066
Grinding:		
Labor	.034	
Steel (balls and liners) ...	.125	
Power	.111	
Miscellaneous	.005	
Total		.275
Treatment:		
Labor	.042	
Chemicals	.126	
Other supplies, repairs, and renewals	.005	
Power	.025	
Total		.198
Precipitation:		
Chemicals	.022	
Repairs and renewals	.007	
Total		.029

	<u>Per ton</u> <u>milled</u>	<u>Total per</u> <u>ton milled</u>
Clean-up and refining:		
Labor	\$0.008	
Supplies	.019	
Marketing bullion	<u>.030</u>	
Total		\$ 0.057
Residue disposal:		
Labor	.011	
Repairs and renewals	.011	
Power	<u>.007</u>	
Total		.029
Water:		
Repairs and renewals	.001	
Power	.018	
Total		.019
Mill, general		<u>.028</u>
Total milling cost		.701